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# The problems of the Silver Dyke Mine

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# T H E  P R O B L E M S  O F  T H E

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# S I L V E R  D Y K E  M I N E

by

D. I. Hayes

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A T H E S I S

submitted to the faculty of the

SCHOOL OF MINES AND METALLURGY OF THE

UNIVERSITY OF MISSOURI

in partial fulfillment of the work required for the

D E G R E E  O F

ENGINEER OF MINES

Rolla, Missouri

1936

Approved by C. R. Forbes
Professor of Mining

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# THE PROBLEMS OF THE SILVER DYKE MINE OPERATION

In a newly-conceived mining enterprise serious operating problems may remain concealed until encountered in the actual early operating period of the property. The major problems of the Silver Dyke Mine were of this nature.

This mine is the property of the Silver Dyke Mining Company, a subsidiary of the American Zinc, Lead and Smelting Company. It is situated about five miles from Neihart, Montana. A three-and-one-half-mile trucking road connects the mine with a railway siding. The property is in the Little Belt Mountains, at an elevation of about seven thousand feet. The climate is severe, the winters long, and the snow fall heavy in the spring.

Prior to equipping the mine for operation, development work in the way of trenching, underground cross-cutting, and churn drilling had demonstrated the existance of an ore-body. Engineers at that time estimated that about 500,000 tons of ore were available, with good prospects of developing an additional tonnage. Sampling had indicated an ore averaging 5.6 ounces of silver; 1.4% lead; and 0.8% copper. Operating costs were estimated as follows:

| | | | |
|---|---|---|---|
| Mining ------------------------ | $1.00 | Per | Ton |
| Milling ----------------------- | 1.27 | " | " |
| General Expense --------------- | 0.35 | " | " |
| State Taxes ------------------- | 0.10 | " | " |
| Or a Total Cost of ------------ | 2.72 | " | " |

The valuable minerals were argentite, galena, chalcopyrite and their oxidation derivatives. The minerals were principally of secondary origin, and occurred in the fractures of a breccia. In the mineralized section, porphyry had intruded the gneiss resulting in an agglomerate consisting of angular fragments of gneiss, of all sizes, within the porphyritic mass. The mineralized rock was intensively fractured, and partly altered by kaolinization. As a result of secondary enrichment, the minerals were concentrated enough near the surface to make an ore. Below the zone of secondary enrichment the primary mineralization was of low grade, and not commercial.

The enriched section ended about one hundred and fifty feet vertically from the surface. The first fifty feet from the surface contained principally oxidized minerals and were lower in metal content than the next one hundred feet, which contained principally sulphide minerals. The ore-body was of tabular shape, with dimensions approximately as follows: One hundred and fifty feet thick, four hundred feet long, and two to three hundred feet wide.

The property was equipped and put into operation in the winter of the year 1923. Operations were permanently discontinued in the spring of 1929, due to ore reserve depletion.

The initial system of mining that was planned and tried was a modification of a system that was used at the Latouche Mine of the Kennecott Copper Corporation. Generally, the method consisted of driving vertical raises at forty-foot centers through the ore block to be mined. These raises were started near the bottom of the ore, and were connected to a grizzly or bulldozing chambers at a lower elevation by inclined finger raises. From the grizzlies other inclined raises connected with loading chutes at the haulage way on a lower level. The ore block was undercut completely at the bottom of the vertical mining raises. From the undercut mining was done upward from the raises by drilling from them forty-five degree down-holes in two rounds from each drilling elevation, which mined out a circular area about thirty-five or forty feet in diameter from each mining raise. The purpose was to mine the entire block as rapidly as possible and draw off currently only the broken ore that was necessary to maintain breaking and filling room under the back of the stope. The completion of the mining would leave a mass of broken ore over the draw points available for drawing when needed. The block of ore developed and mined by this system was eighty feet wide,two hundred feet long, and one hundred fifty feet thick. This ore block was developed and broken successfully at a cost of about eighty thousand dollars.

The mining of the vertical raises in this block was completed to a point near the surface early in the spring, before the winter breakup and thaw. Soon a heavy accumulation of snow began to melt, and a great amount of surface and underground water ran into the mass of broken ore, and found its way to the draw points at the bulldozing grizzlies. Considerable pressure developed at the draw points, and had to be relieved by drawing ore, as the timbers would not stand the weight. At this time, it was discovered that the rock in which the ore occurred broke down and disintergrated very rapidly. Under wet and aerating conditions, a fragment of the rock, which was hard, crystalline, and abrasive when freshly broken, would change in a few weeks to a soft, clayey mass, which could be crushed by stepping on it. This condition, together with the water, soon developed a stope full of semi-mud or fluid, instead of broken crystalline rock, as was originally anticipated. The broken ore, in this condition, ran freely so long as it was sufficiently wet, but became quite sticky, and refused to run at all when the water sources dried up as the season advanced. Water was supplied to the draw points and chutes in order to get the ore through the grizzlies and loaded into the cars. In a short time, the draw points above the grizzlies drew through to day-light at the surface in the form of a vertical, round hole five or six feet in diameter. Further undercutting of the broken ore near the back of the grizzlies would not promote caving. Wind shooting in the open draw points was tried without the desired results.

In order to get production, it was necessary to start re-mining the ore from the surface at the top of the holed-through draw points. This was done in the beginning by the use of picks and bars, but the ore was so sticky that this method was too slow and costly. Finally, it was demonstrated that drilling and blasting again gave the best results. The broken ore mass had solidified sufficiently so that it could be drilled with an air drill, if air were continuously blown through the hollow drill-steel while drilling.

The final result was that the block of ground that had first been mined from the bottom up had to be re-mined from the top down. The first mining did not facilitate the second mining, since mining from the top down would have been less costly had the ground been in place so that it could have been freshly broken as needed.

The re-mining of this block from the top down was started in the early summer, and continued throughout the following winter and spring. As soon as frosty and freezing weather set in, the open-top work became more difficult and costly. It was found that the frosty ore hung up in the draw points and chutes. Unless the plank-lined chutes were cleaned out daily by hand, the fine, frosty ore would accumulate on all sides until the opening became completely closed. After the heavy snows started, the ore-handling difficulties were increased greatly, and at times it was impossible to get any appreciable tonnage from the mine.

During this period of mining difficulties it was demonstrated that a more satisfactory mining method had to be devised. The cycle of difficulties experienced indicated to a considerable extent the conditions to be contended with, which could not be controlled. They were as follows:

(a) A sticky ore, which packed when broken, and became consolidated in time, until it was self-arching, and would support itself almost as well as the unbroken ore in place;

(b) A climatic condition that would not permit satisfactory open-top mining for periods varying between five and seven months each year;

(c) A wet and dry season within the ore-body.

The partly oxidized condition of the ore made it desirable to mine by vertical slicing, if possible, in order to produce currently a good, average mix of oxidized and sulphide ore. Safety and low cost were necessary requirements. The limit of cost was about $1.00 per ton, including development, and all other charges properly chargeable to mining.

A method of vertical slicing over a draw point and around raises was conceived, and development work for it was done during the first winter's mining. This system was put into operation the following spring with considerable success.

Two double, loading pockets were established above a haulage tunnel, and one hundred feet apart. The top of each pocket supported a grizzly about forty feet long, which was provided with the necessary manways. At each end of the two grizzlies, or bulldozing chambers, raises were driven to the surface. This located four vertical, five-by-five-foot raises at the corners of a rectangle forty feet wide and one hundred feet long. The raises were approximately one hundred seventy-five feet long, and were timbered with six-by-six-inch wall plates spaced at five-foot centers, and equipped with hanging rods, so that when a raise was holed through, the entire timbering structure could be supported and suspended from bearing sills at the surface.

The raises were undercut, and the draw points from the grizzlies widened. The undercutting between the raises over the same grizzly covered an area about twenty feet wide and fifty feet long, and sloped upward at about forty-five degrees from each draw point. The raises were reamed to about fifteen feet in diameter to a point within forty feet of the surface. The reaming was done from the bottom up,and was accomplished by drilling a ring of forty-five degree downholes off each five-foot set of wall plates. The blasting of the holes was done a ring at a time from each successive set of wall plates, which were hung by rods from the top for that particular purpose.

During the period that the underground work was in progress on this block of ground, two churn drills were drilling blast holes from the surface. The first holes were spaced about eight feet outside the perimeter of the reamed raises, and about the same distance apart. The burden spacing of the holes was increased as the distance away from the center of the raises increased until a maximum burden of about fifteen feet was reached. The blast holes were drilled with a five-and-five-eighths-inch bit. An effort was made to keep the holes straight to avoid irregular blasting burdens as much as possible.

The holes were loaded with explosive charges of variable lengths spaced with intervals of tamping. A typical charge was three or four feet of powder, alternating with eight or ten feet of tamping for the entire vertical distance along the portion of the hole to be broken. The detonation of these spaced charges was handled by the use of Cordeau Bickford detonating fuse. The total charge was calculated on a tonnage basis, and the spacing of the explosive was made to conform to the vertical distance to be broken and the apparent burden at the different elevations. All of the first group of holes were loaded and shot with the idea of not caving the surface. The purpose of preserving the surface, in this case, was not a necessity, but it was desirable for experimental purposes. It was demonstrated that the holes, if loaded properly, would slice vertically as desired, and result in very little back breaking or disturbance at the collar of the hole. It was indicated that the bottom half of a hole could be blasted first, and the top half loaded and shot later, if this procedure were desirable to prevent too much accumulation of broken ore.

This system of mining worked out very well. Difficulties of drawing the ore were encountered even when the ore was shot down in parcels of two thousand tons or less. The blast hole shooting eventually connected the four raises, and a wide opening was shot out underground before the surface over this opening started to cave. Eventually an open-top glory hole resulted, with more or less vertical walls from the surface down to the angle of repose of the broken muck in the bottom of the pit. When the blast hole shooting had been completed, a round pit about one hundred and fifty feet in diameter had been developed. On account of the sticky nature of the ore, a considerable tonnage remained in the bottom of the pit. This was because it would not run to the draw points. This accumulation was cleaned out by hand work safely, as very little sluffing occurred from the vertical walls.

The initial experience with blast-hole mining indicated that vertical slicing could be accomplished by blast holes, and that they could be loaded and shot in such a way as to allow the surface to remain intact as a surface pillar for a period of time that was dependent on the span between the wall supports and its vertical thickness. The surface pillar formed an arched back over the stope opening, with weight developing where the back joined the vertical walls. From time to time these corners would fail, and then caving would be promoted in the form of peeling until a new secure arch line was reached at a point a short distance higher in elevation. This caused material to fall out of the back frequently for periods of two or three days at a time, after which very little, if any, caving would occur for a week or two.

In the fall, after the blast-hole method had demonstrated itself, development work was started on two blocks of ground for the following winter's mining. These blocks were laid out with the idea of maintaining a roof or surface pillar over the blast-hole stopes throughout the winter period to protect the mining operation against the difficulties formerly experienced with open-top mining during the winter.

The two blocks were thirty feet wide and one hundred feet long, and spaced sixty-five feet apart, center to center, which provided for a pillar thirty-five feet wide between them. The plan was to mine the two thirty foot by one-hundred-foot blocks during the winter and maintain the roof pillars. In the spring, after the roof pillars had caved, it was contemplated to mine the pillar between the two winter blocks into the then open-top glory hole. The roof pillars were maintained as a weather protection for about four months until they gradually caved through to the surface. It was demonstrated again that a roof pillar could be sustained for a period of time depending upon its thickness, as it gradually sluffed away and did not fall as a block or a large mass.

Throughout the life of the operation these mining methods were used with modifications. The principal modifications were closer-spaced draw points,and the blast hole slicing supplemented in part by the caving of the walls, which was promoted by undercutting them from retreat manways where the surface conditions did not permit of the drilling of blast holes.

The economics of the mining method as it was developed is best illustrated by the following tabulated cost comparsions:

| Year | Tons | Mining Cost | Development Cost | Total Mining Cost |
|---|---|---|---|---|
| 1923 | 106,076 | $0.65 | $0.30 | $0.95 |
| 1924 | 148,664 | 0.54 | 0.30 | 0.84 |
| 1925 | 167,847 | 0.40 | 0.30 | 0.70 |
| 1926 | 223,924 | 0.39 | 0.30 | 0.69 |
| 1927 | 263,868 | 0.30 | 0.30 | 0.60 |
| 1928 | 192,583 | 0.407 | 0.226 | 0.633 |

Original mill tests preliminary to mill design and construction recognized that the ore as mined would be partly oxidized. A typical mill test indicated a recovery of 84.02% of the silver; 86.69% of the lead and 87.47% of the copper on a head sample assaying 5.4 ounces silver; 1.2% lead; and 0.77% copper.

The mill was designed and built with a flow sheet that provided for the tabling of the flotation tails in order to recover as much oxidized mineral as possible from the ore. From the beginning the mill recoveries were disappointing, principally for the reason that the degree of oxidation of the ore and its physical characteristics were not fully anticipated. The first year's mill recoveries averaged 61.80% of the silver; 40.70% of the lead; and 51.65% of the copper on a mill head of 4.97 ounces silver; 1.62% lead and 0.75% copper.

A considerable amount of experimental work was done in the laboratory in an effort to work out improvement in the flotation recoveries. Various sulphidizing schemes were tried, and encouraging results were obtained in the laboratory. Two sulphidizing methods were tried in the plant, one using calcium sulphide, and one using sodium sulphide. Neither of the plant tests on sulphidizing gave satisfactory results.

In an effort to increase recoveries, it was decided to reverse the original scheme, and table ahead of flotation. Another ball mill was installed to permit of relatively coarse grinding in the primary mill for a table feed, and then regrinding the table tailings for the flotation feed. This re-arrangement did not obtain the results expected, and it became apparent that another serious interference existed, which was an unusually large proportion of colloidal slime. The average ore contained from fifteen to twenty per cent colloidal slime, which was naturally coagulated, causing the mill pulp, even when coarsly ground, to be quite thick when it contained as low as 20% solids. It had also become apparent that this colloidal slime condition was a serious hindrance to both gravity and flotation concentration.

Experimental work indicated that the ore contained ferrous sulphate, which caused the colloidal material to coagulate. Lime that was used in the mill circuit destroyed the ferrous sulphate, as such, but the lime was also a coagulent of the colloids. A number of chemicals were found that would disperse the colloids completely, and change the average pulp from a thick fluid to a normal, thin, free-settling condition at ordinary dilutions. A limited amount of water available made it necessary to settle the mill tailings and recover the maximum amount of water for re-use. In order to obtain a return of clear water it was necessary that the colloidal slimes be coagulated during the settling process, otherwise, clear water would not develop on the settling tanks.

It was found that barium sulphide would disperse the pulp when not added in sufficient amount to cause the pulp to become alkaline. When the pulp became alkaline with barium sulphide, the colloids coagulated very quickly. The addition of lime to a pulp that had been dispersed with barium sulphide caused normal coagulation of the slime, and caused clear water to develop on the settling tanks for return to the system. The amount of barium sulphide required varied between one and four pounds per ton. This reagent was bought as a crude product known as black ash, and cost about 2½¢ a pound.

The flow sheet of the mill was changed to permit of de-sliming by use of barium sulphide. Two complete and independent mill circuits were now operated, one called the "slime circuit" and the other the "granular circuit". The mill feed was de-slimed after being ground to about ten mesh. The slime portion was sent direct to the slime circuit flotation machines. The granular material was classified and tabled. The table tails were re-ground and delivered to the granular circuit flotation machines.

The improvement in results after the colloidal material was separated from the ore was marked. The granular part of the mill feed, consisting of about 80% of the total, became more nearly a normal ore, as the slime developed in the regrind for flotation was not excessive. Flotation recoveries in the slime circuit were very low. Laboratory experimental work indicated that near normal flotation recoveries could be obtained from these slimes. The laboratory results could not be duplicated in the plant, due to a lack of flotation capacity. For experimental purposes a fifty-ton-capacity flotation machine was installed. This machine could be run continuously by feeding it with any amount of slimes cut out of the regular mill circuit. It was found that in order to get flotation recoveries comparable with recoveries obtained in the granular circuit, the feed had to be reduced to a rate of about five tons per twenty-four hours for the fifty-ton machine, and the dilution maintained at about 7 or 8% solids. For economic reasons, it did not seem advisable to provide additional flotation capacity for this small amount of feed.

The results obtained from de-sliming were a considerable improvement over previous results when de-sliming was not used. This may be illustrated by the following figures, representing a year's average on a mill head assaying 4.44 ounces silver; 1.54% lead; and 0.76% copper:

| | Silver | Lead | Copper |
|---|---|---|---|
| Recoveries: | 82.77% | 73.48% | 72.95% |

After three years of operation, it became apparent that costs would have to be reduced, as the ore was becoming lower grade and more oxidized. Operating costs for period 1923 to 1928, inclusive, were:

| Year | Tonnage | Mining Cost | Milling Cost | Other Costs | Total Cost |
|---|---|---|---|---|---|
| 1923 | 106,076 | 0.95 | 1.44 | 0.43 | 2.82 |
| 1924 | 148,664 | .84 | 1.27 | 0.41 | 2.42 |
| 1925 | 167,847 | .70 | 1.50 | 0.40 | 2.60 |
| 1926 | 223,924 | .69 | 1.10 | 0.28 | 2.07 |
| 1927 | 263,868 | .60 | .83 | 0.15 | 1.58 |
| 1928 | 192,202 | .633 | .726 | 0.117 | 1.476 |

The mine was being operated at a profit, but profits were smaller than had been anticipated. Considerable prospecting had been done without success in showing new ore extensions. The ore in sight was, therefore, definitely limited, and a salvaging policy was necessarily adopted. Milling and general overhead costs were high, as compared with mining costs. The best opportunity for reducing costs seemed to be in the milling department. Additional tonnage seemed to be the answer, if this could be accomplished without heavy capital expenditures.

As previously explained, the ore was a secondary mineralization. The mineral occurred almost entirely as a fracture filling in a coarse breccia. The idea was conceived of getting rid of waste rock after primary crushing. It was found that a large proportion of the waste could be picked off the conveyor belt, which was transporting ore crushed to four inches. Screening tests were made on the primary crushed ore, and it was demonstrated that the rock that remained on a 2-inch square opening screen carried but a small portion of the total mineral, and that the mineralized pieces were few and conspicuous, and could be hand-sorted readily. A vibrating screen was installed providing for the delivery of the oversized to a picking belt, which discharged into a waste disposal bin. Two men per shift could easily pick from this belt most of the mineralized rock. By this means about 30% of the ore mined was disposed of as low-grade material. During the year 1927, 79,285 tons were wasted, which assayed approximately 1.35 ounces silver; 0.51% lead; and 0.32% copper. This compared with a flotation mill tail assay of 1.10 ounce silver; 0.44% lead; and 0.32% copper. It is apparent that the wasting method increased metallurgical losses, but reduced operating cost, and increased the profit margin per ton of ore mined. The plant capacity was increased about 43%, and the over-all cost per ton reduced about 49¢ per ton of ore mined, or about 24.00%.

The disposal of the waste after removal from the mill circuit was a difficult problem. It was thought that a mechanical method of waste elimination would be more desirable, both from the standpoint of waste disposal and a minimum mineral loss. A five-cell 30x36 inch Joplin type cooley jig was built for experimental purposes. It was found that when this jig treated material sized to minus one-half inch and plus .065 inch, a very low-grade tailing could be discharged when making a low-grade concentrate that contained considerably more mineral than the general mill feed after the plus .065 inch portion had been screened out.

Experimental results justified the installation of the jig wasting system in the plant. A cone crusher was installed so that the whole mill feed could be crushed to one-half inch size. Two five-cell 36x42 inch jigs were built and accessories installed. This new flow sheet eliminated the necessity of the use of one of the two ball mills.

The operating results were quite satisfactory. Accurate means were not available to determine the amount of material disposed of as a jig tailing, but the proportion was known to exceed 45%. This method of treatment increased mill capacity about 20%. The mill that had originally been designed for about 400 tons per day now would handle about 900 to 1000 tons a day, with no enlargement of the original floor space.

During the year 1928, this method of wasting produced the following results:

| | Silver | Lead | Copper |
|---|---|---|---|
| Jig tails | 1.58 Oz. | 0.38% | 0.30% |
| Flotation tails | 1.53 Oz. | 0.65% | 0.36% |

The above tailing assays were for the year's average. The average ore treated contained a high percentage of oxidized mineral. During one month when the oxidized mineral content was low, the tailing average was as follows:

| | Silver | Lead | Copper |
|---|---|---|---|
| Jig tails | 0.73 Oz. | 0.16% | 0.17% |
| Flotation tails | 0.72 Oz. | 0.24% | 0.15% |

By use of this system of milling, the total per ton operating cost was reduced from $1.58 per ton in 1927 to $1.476 per ton in 1928, including the write-off of the cost of the new equipment installed.

The ore reserves became depleted in the spring of 1929, and the operation was discontinued. The writer became associated with this operation in 1923, four months after operations began, as Assistant Manager, and became Manager early in 1925, remaining until the operations were discontinued.